Technical Paper 41 — 55

UNIVERSITY LIBRARY
DEC 17 1913
PRINCETON, N. J.

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

MINING AND TREATMENT OF LEAD AND ZINC ORES

IN THE

JOPLIN DISTRICT, MISSOURI

A PRELIMINARY REPORT

BY

CLARENCE A. WRIGHT

WASHINGTON
GOVERNMENT PRINTING OFFICE
1913

Generated on 2024-09-28 13:56 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

First edition. June, 1913.

Generated on 2024-09-28 13:53 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google

Digitized by Google

Original from
PRINCETON UNIVERSITY

CONTENTS.

	Page.
Introduction	5
Acknowledgments	6
General nature of mining and milling in the district	6
Location of the Joplin district	7
Production	8
Geology	9
Ore deposits	9
Occurrence	9
Mineral constituents	10
Mining methods	11
Underground work	11
Shafts	13
Drilling	14
Blasting	15
Shoveling	17
Haulage	18
Timbering	18
Pumping	19
Surface work	19
Hoisting	19
Surface tramming and storage of ore	20
Power	21
Ore-dressing methods	21
Concentrating mills	23
Coarse concentration	23
Sorting	23
Crushing	26
Jigging	27
Concentration of fine material	29
Table work	30
Handling and character of the concentrates	32
Source and quantity of waste	33
Mill tests	33
Labor	36
Accidents and safety precautions	37
Health conditions at the mines	39
Conclusion	40
Publications on mine accidents and methods of mining	42

(RECAP)

476786

Generated on 2024-09-28 13:56 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google

Digitized by Google

Original from PRINCETON UNIVERSITY

ILLUSTRATIONS.

		Page.
FIGURE 1.	Underhand stoping as practiced in the Joplin district	11
2.	Plan of "sheet-ground" mine in the Joplin district	12
3.	Three ways of inserting detonator in primer	17
4.	Arrangement of an average zinc concentrating mill in the Joplin district	22
5.	Flow sheet of large zinc mill in the Joplin district	24

Generated on 2024-09-28 13:55 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

MINING AND TREATMENT OF LEAD AND ZINC ORES IN THE JOPLIN DISTRICT, MISSOURI; A PRELIMINARY REPORT.

By CLARENCE A. WRIGHT.

INTRODUCTION.

This paper aims to outline the general conditions affecting the safety and efficiency of the methods used at the lead and zinc mines of the Joplin district, Missouri, in the mining and treatment of the ores, and also to indicate certain possible improvements in present mining and milling practice. It does not attempt to describe, except incidentally, the geology of the district, as this has been discussed in detail by W. S. T. Smith and C. E. Siebenthal [a] and by H. F. Bain [b] in reports published by the United States Geological Survey, and also by E. R. Buckley and H. A. Buehler in reports of the Missouri Bureau of Geology and Mines.[c]

The area covered by this paper does not include all the camps of the district, but only the more important mines of the camps visited by the writer. Other important camps, such as Miami, Spring City, and Granby, would have been visited if time had permitted. A subsequent report, giving a more detailed discussion and covering more extended field researches, is planned. The present paper is a preliminary statement dealing with the more important features of mining and milling methods and the factors that govern safety and efficiency. It is the first of a series of reports on these features of the lead and zinc industry of the United States, and is published by the Bureau of Mines in the hope that it may aid in the development of safer and more efficient mining and milling methods in that industry.

Although the lead ores are closely associated with the ores of zinc, the proportion of lead to zinc at present is far less than in former years. The zinc ores are here considered more especially because the losses in their milling are proportionately greater than in the case of lead ores. Only the mines actually examined and tested are considered in detail, statements of results being restricted to the mines in general, with no special reference to any one mine. Most of the

[a] Joplin district folio (No. 148), Geol. Atlas, U. S. Geol. Survey, 1907.
[b] Twenty-second Ann. Rept. U. S. Geol. Survey, pt. 2, 1901, pp. 111-215.
[c] Geology of the Granby area: Mo. Bureau of Geology and Mines, vol. 4, 2d series.

Generated on 2024-09-28 14:03 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

tests in the mills were made with a view to discovering where the losses were greatest rather than to determining the total metal recovery, for an accurate mill test was almost impossible at any of the concentrating plants in the district, either because of the tonnage of ore treated being estimated from the number of tubs or cars hoisted in a day, or in many cases because of the poor arrangement of the mill for making tests.

ACKNOWLEDGMENTS.

The writer is indebted to H. A. Buehler, State geologist of Missouri, who offered many suggestions; to C. E. Siebenthal, of the United States Geological Survey; and to the operators and mine superintendents who furnished information and extended courtesies.

GENERAL NATURE OF MINING AND MILLING IN THE DISTRICT.

Although the mining and milling in the Joplin district may seem crude and wasteful to those accustomed to more elaborate methods used elsewhere, yet many skilled mining men of wide experience and successful in other districts have made mistakes in their first attempts here. The ore bodies are exceedingly variable in shape, and for this reason an attempt to block out the ore is seldom made, as it is difficult to determine the amount an ore body will yield from the records of the drill holes. It is often the case, however, that the amount of drilling and preceding development work is too limited. In fact, most of the failures here have been due to insufficient prospecting, shafts having been sunk and mills erected without any certain knowledge of the extent of the underlying deposits. This is especially true of the ore bodies found in other than blanket or "sheet-ground" deposits.

Characteristic of this district are the simple methods used and the low cost of equipment. The building of small and inexpensive plants has been largely due to the general small size and irregular and uncertain nature of the ore bodies. The loss of time incident to failure to furnish a continuous supply of ore is more costly with a large mill than with a small one, so that large central mills have not been considered feasible in this district. Economy in first cost, rather than a higher saving by more efficient equipment, seems to be the ruling consideration among the operators. Increased capacity at small cost is sought, a result usually essential at mines where the ore bodies are as low grade as they are here. The small size and the low grade of most of the ore deposits, as well as the high royalties demanded, naturally lead to conservative investments.

In spite of these conditions, however, the methods used in both mining and milling are constantly improving. The recovery of the

Generated on 2024-09-28 13:58 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

minerals from the ores in the milling practice and the conditions at the mines are being bettered each year. This result has been due largely to the higher prices received for the concentrates, which have made possible the profitable working of much thinner deposits than formerly, and partly to the possibility of making a closer saving in the milling of these lower grade ores through these better prices. A few years ago ores with less than 4 per cent recoverable mineral content were considered unworkable, but now many of the "sheet-ground" deposits of ores with a recoverable mineral content of less than 3 per cent are being worked profitably.

LOCATION OF THE JOPLIN DISTRICT.

Few mining districts have as many mines and mills in an area as small as that of the Joplin district, better known as the southwest Missouri lead and zinc district. Although zinc-ore deposits are found in other parts of the State, the most important are included in the Joplin district, which comprises the southwestern part of Missouri, the northeastern part of Oklahoma, and the southeastern part of Kansas. The area is divided into separate mining camps, but the greater proportion of the mines is found in Jasper County, Mo., embracing the camps[a] of Joplin, Webb City-Carterville, Oronogo, Carl Junction, Carthage, Alba-Neck City, Duenweg, and Cave Springs. Some of the other more important camps are the Miami in Oklahoma, the Galena in Kansas, and those in Missouri near Granby, Newton County, and Aurora, Lawrence County. Taking the largest producing area of the district, however, and Joplin as the center point, nearly all of the camps could be included within a radius of 15 miles.

The names and locations of the mines covered in this report are as follows:

Camps and mines covered in this report.

Name of mine.	Section.	Location.		Name of mine.	Section.	Location.	
		Township.	Range.			Township.	Range.
ALBA-NECK CITY CAMP.				CAVE SPRINGS CAMP.			
Jasper County, Mo.				*Jasper County, Mo.*			
Cliffwood	1	29	33 W.				
Little Mary	1	29	33 W.				
Quick Seven	6	29	32 W.	Kramer	2	27	34 W.
CARL JUNCTION CAMP.				DUENWEG CAMP.			
Jasper County, Mo.				*Jasper County, Mo.*			
Molar-Smith	7	28	33 W.				
United Lehigh	7	28	33 W.				
Weaver	17	28	33 W.	Carmean-Squires	34	28	32 W.
				Coahuilla	33	28	32 W.
CARTHAGE CAMP.				Durby	34	28	32 W.
Jasper County, Mo.				Etta B	34	28	32 W.
				Nowata No. 1	28	28	32 W.
S. V. and D	31	29	34 W.	Vogey (American)	34	28	32 W.

[a] The word camp as here used signifies a group of mines and the adjoining settlement, whether a village or city.

Generated on 2024-09-28 13:54 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

Camps and mines covered in this report—Continued.

Name of mine.	Section.	Location.	
		Township.	Range.
GALENA CAMP.			
Cherokee County, Kans.			
Diplomat	23	34	25 E.
Peacock Galena	14	34	25 E.
JOPLIN CAMP.			
Jasper County, Mo.			
A. B. C. Mining Co	32	28	33 W.
Amalgamated	4	27	33 W.
Bay State	4	27	33 W.
Grand Haven	32	28	33 W.
John Jackson	32	28	33 W.
Joplin Kentucky	32	28	33 W.
Mary "C"	16	28	33 W.
Prairie Chicken Mining Co	3	27	33 W.
Red Lion Mining Co	4	27	33 W.
Samson Mining Co	6	27	33 W.
Sitting Bull	21	28	33 W.
Newton County, Mo.			
Mattes Bros	24	27	33 W.
Jackson Zinc & Lead Co	24	27	33 W.
ORONOGO CAMP.			
Jasper County, Mo.			
Oronogo Circle	36	29	33 W.
Pinnacle	36	29	33 W.
WEBB CITY-CARTERVILLE CAMP.			
Jasper County, Mo.			
Albert F	28	28	32 W.
American Davey No. 1	21	28	32 W.
American Davey No. 2	21	28	32 W.
American Davey No. 3	21	28	32 W.
American Davey No. 4	21	28	32 W.
Athletic	12	28	33 W.
Bohemian Girl	7	28	32 W.
Continental Zinc	21	28	32 W.
Culpeper	8	28	32 W.
Disbrow	17	28	32 W.
Edmund Lyon	12	28	33 W.
Empress	17	28	32 W.
Endeavor	21	28	32 W.
Good	12	28	33 W.
Ice Plant	7	28	32 W.
John "L"	12	28	33 W.
M. S	28	28	32 W.
Mercantile No. 4	7	28	32 W.
Mercantile No. 5	7	28	32 W.
Mineral King	7	28	32 W.
Missouri Mule	7	28	32 W.
Osceola	7	28	32 W.
Providence	7	28	32 W.
Ramage	17	28	32 W.
Schoenherr-Walton	8	28	32 W.
Sunflower	8	28	32 W.
Yellow Dog	6	28	32 W.

The two most important camps that have been opened in recent years are Miami, Okla., and Thoms Station, a short distance northwest of Joplin, Mo. The deposits of the latter are in a "soft-ground" formation and are considerably richer in zinc content than the general average of the mines in the other camps of the district. Being located near Joplin, the Thoms Station mines are included in the foregoing tabulation under the Joplin camp.

Nearly all the camps are accessible by railroads and fairly good country roads, so that the shipment of ore from most of the mines is an easy matter.

PRODUCTION.

The Joplin district is one of the main ore-producing areas of the Ozark region.[a] It is not only the principal zinc-producing district of this region, but more spelter is produced from its zinc ores than from those of any other district in the United States. According to the report given by the United States Geological Survey in Mineral Resources of the United States for 1910, of 303,139[b] short tons of zinc produced in the United States 140,653 tons were produced from

[a] The Ozark region is a dissected plateau, occupying most of the southern half of Missouri and parts of Kansas, Oklahoma, and Arkansas. In it and around its border are deposits of lead and zinc ore of great commercial importance.

[b] These totals are based on smelter reports.

Generated on 2024-09-28 14:05 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

the zinc ores of the State of Missouri, the Joplin district being the chief contributor. The production from the district as a whole has remained about the same for the past two or three years. It is probable that with more prospecting new ore bodies will be discovered, and these will help to uphold the high production that the district has maintained in the past.

GEOLOGY.

According to the United States Geological Survey the rocks in which the ore bodies of the Joplin district occur consist mainly of sedimentary beds which lie within the Mississippian and overlying Pennsylvanian series, of Carboniferous age. The greater proportion of these deposits occur in the Boone chert of the Mississippian series, but at Miami, Okla., important ore bodies are found not only in the Boone chert but in the overlying rocks of the Chester group. These formations consist of limestones, cherts, and dolomites, with lesser quantities of shale and "soapstone." The geological succession of the rocks is simple, and the formations, though nearly flat, have a general dip to the northwest. Relatively speaking, there has been little disturbance throughout this area either by faulting or lateral movement, the most noticeable disturbances or unconformities being between the Pennsylvanian and Mississippian formations.

ORE DEPOSITS.

OCCURRENCE.

The lead and zinc ores are distributed throughout this area in rather well-defined blanket (better known as "sheet-ground") deposits, or in "runs" or irregular deposits.

The "sheet-ground" deposits include those ore bodies that lie between the stratified layers of bedded cherts and are in a nearly horizontal position. They vary in thickness from a fraction of an inch to several inches, and here and there contain cavities lined with crystals of lead and zinc minerals with minor amounts of iron sulphides and other minerals. The ore beds are 6 to 20 feet thick and are far more uniform in richness of mineral content and are more reliable in lateral extent than the other forms of deposits. Brecciation is relatively unimportant in these "sheet-ground" deposits.

The "runs" and irregular bodies may be divided into two classes—those in fairly hard ground where little timbering is necessary and those known as "soft-ground" deposits, which require timbering in nearly all the drifts as mining advances. These ore bodies are usually of an elongated form and at times are of considerable length and thickness. A few are circular or semicircular in shape and are known as "circle" deposits. The so-called "runs" and the irregular

Generated on 2024-09-28 13:53 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

ore bodies are found at shallower depths than deposits of the "sheet-ground" type and are as a rule considerably richer in mineral content, often containing large pockets of nearly pure blende and galena. These minerals are commonly found associated with a dark-colored secondary chert as the main filling material of fractures and cavities.

The most important gangues in which zinc blende and galena occur are chert and dolomite with minor amounts of limestone, shale, soapstone, and mud. In some of the deposits the chert and dolomite are intimately mixed, the zinc blende being embedded in a black secondary chert, which serves to cement the gangue with the blende. Chert or flint is the characteristic gangue material of the ore deposits in the district, and can be recognized in three different varieties, the white chert, the blue to gray chert, and the black or secondary chert. The blue chert is known as "live flint" and its occurrence is usually an indication of the presence of blende. The black variety, which has already been mentioned as filling fractures, cavities, and the interstices of the brecciated rock, is generally found in the brecciated areas closely associated with sphalerite, or in places with galena, and as a cementing material or matrix for the blue and white cherts with zinc blende. The so called "cotton-rock," which is a decomposed chert, is often found in layers several inches thick near the surface or overlying the ore bodies. This dead-white flint or "cotton rock" is noticeable in the roof of some of the "sheet-ground" mines, where it usually has to be taken down as waste along with the ore or has to be propped up with posts at frequent intervals.

As regards the vertical distribution of the ores mined, the oxidized ores of zinc and lead generally occur above and near the level of groundwater; below this level the sulphide of zinc becomes dominant, the proportion of galena becoming less and that of iron sulphides greater as the depth increases. The zinc blende is seldom found in the weathered zone for it becomes oxidized, and generally takes the form of silicate or carbonate of zinc, as the result of the action of surface waters. As the general course or surface waters is downward, and as the blende is more soluble and more easily oxidized than galena, it is also carried downward and is found at a greater depth below the influence of the oxidizing agents. The probability, therefore, is that where a large body of galena is found with scattered amounts of blende, there will be a good deposit of blende below it.

MINERAL CONSTITUENTS.

The most important minerals of commercial value occurring in the district are galena, cerussite, sphalerite, calamine, and smithsonite. These minerals are better known locally as "lead," for galena, "blende" or "rosinjack," "rubyjack," "steeljack," or "blackjack" for sphalerite; calamine and smithsonite are classed together under one term

Generated on 2024-09-28 13:54 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

as "silicate." Closely associated with these minerals, are pyrite, marcasite, chalcopyrite, calcite, dolomite, and, in a few mines, small crystals of quartz. These minerals also have their local terms such as "mundic" for the iron sulphides, "tiff" for calcite, and "spar" for dolomite. Besides these, in a few of the mines there are small deposits of the hydrocarbons, which are termed by the miners "coal tar" or "asphalt." This "tar" or "coal tar" occurs in small openings or crevices and is usually viscous at the ordinary temperature of the mines. A yellow to light-brown clay also occurs, especially in the "soft-ground" deposits, and, although it is not a definite mineral, it in many places contains considerable zinc silicate. Several other minerals occur in the deposits throughout the district, but are less common and important.

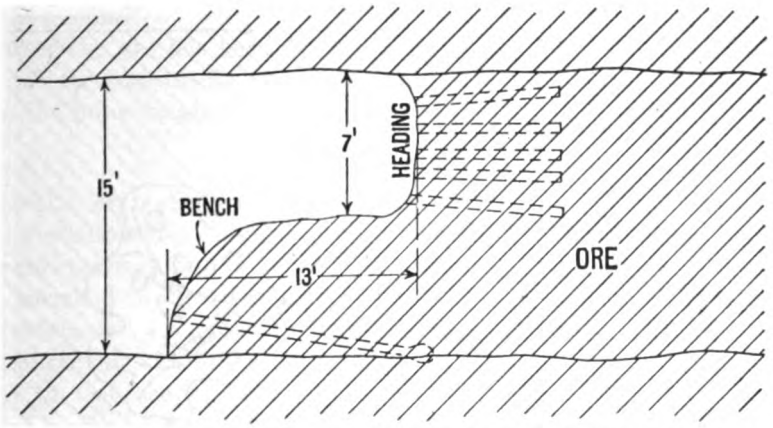

FIGURE 1.—Underhand stoping as practiced in the Joplin district.

MINING METHODS.

UNDERGROUND WORK.

The methods of mining employed in the extraction of the ore from these deposits vary with the nature of the deposits and have been developed in the mines to suit the varying conditions encountered. The surrounding topography is relatively flat, so that the mining is done from vertical shafts, 100 to 300 feet deep, sunk at frequent intervals. In general the ore bodies are mined from one level, as the deposits themselves are usually flat and seldom more than 20 to 30 feet thick.

Where possible, a system of underhand stoping (fig. 1) is practiced in the district, especially in the "sheet-ground" mines, if the face of the ore is high enough to warrant it. A machine drill is set on a 7-foot column placed at the upper part of the face near the roof in order to advance a heading about 6 to 8 feet high, leaving a bench

Generated on 2024-09-28 14:09 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

beneath of 8 to 12 feet thick above the floor of the stope. When the heading or breast has been advanced 15 to 20 feet, the bench is drilled and blasted. In many deposits only the upper layers of the "sheet ground" are rich enough to be worked profitably; in other deposits only the lower layers can be worked with profit. The chief cause of this condition is the presence of an intermediate layer of barren ground, 3 to 6 feet thick, which has to be mined with the ore. Where stoping is not done, there seems to be no special system of mining except that of following the ore bodies, pillars being left wherever necessary.

Where the deposits are in the nature of "runs" or irregular bodies, they are often large enough to be worked from more than one level. In a few of the mines they are so worked, but the usual method is to follow the orebodies to their upper limits by raises cut through from

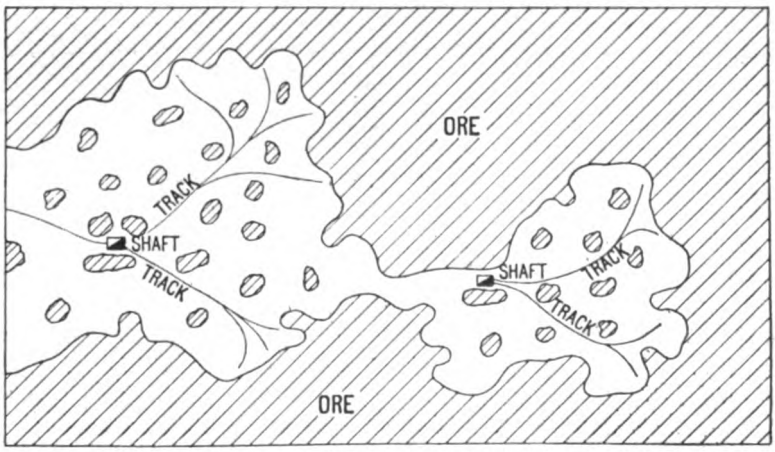

FIGURE 2.—Plan of "sheet-ground" mine in the Joplin district.

the lower levels, the broken ore being allowed to fall down along a slope or chute and work itself to the bottom, where it can be shoveled into the cars or tubs. When the breast of these raises has been driven far enough ahead, the bench formed is underdrilled at the bottom and blasted, high walls on each side being left for the support of the roof. This system is continued and results in drifts or cuts 40 to 60 feet high and 20 to 40 feet wide.

Except in those mines where "soft-ground" deposits occur, little timbering is done, though in the "hard-ground" mines occasional props or posts are necessary to support slabs and to prevent the rock from scaling off. In place of timbering, pillars are left at intervals of 25 to 50 feet, and vary in thickness from 12 to 30 feet, depending on the nature of the deposits and the firmness of the overlying strata (fig. 2). The average thickness of the pillars left to protect the shafts is about 40 feet.

Generated on 2024-09-28 14:01 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

The roof of the "sheet-ground" mines usually consists of a layer of hard flint, averaging 2 to 3 feet thick, which serves both as a support for the overlying beds and as a clean "back" for the men to work against. Where this flint layer is missing, the roof has to be trimmed after each round of shots. For this purpose most of the mines employ special men known as roof trimmers. As previously mentioned, in some of the mines a thin layer of white "cotton rock," 2 inches to 1 foot thick, occurs in the roof. This cotton rock is a decomposed chert and in places is soft and can hardly be supported by posts, so that it has to be taken down and shoveled into the tubs with the ore. This foreign material, of course, increases the amount of "dead rock" to be handled and decreases the richness of the ore as a whole.

The proportion of the deposit left as pillars depends upon the nature of the ground.

After the ore has been broken it is shoveled into cars and hauled on narrow-gage tracks to the shafts, where it is hoisted and dumped into the hoppers above.

SHAFTS.

As the ore bodies that are being mined at present lie at comparatively shallow depths, the sinking of shafts is an easy matter. With the exception of two or three mines that have inclines, the ore bodies are reached by vertical shafts that have as a rule only a single compartment, except at some of the larger properties, where double-compartment shafts are to be found. Most of the shafts are sunk 5 to 12 feet deeper than the levels from which the ore bodies are being worked, so as to form a sump. Over this sump a platform is built for handling the tubs and ore cars. From the collar down, the upper part of the shaft is cribbed to stop any loose ground or bowlders from falling. This cribbing is extended through the alluvium and less firm ground until hard rock is reached, the depth depending upon the nature of the ground and the amount of water coming from the upper beds. Only a few shafts were noticed where the cribbing extended nearly to the bottom of the shaft, and these were sunk in "soft-ground" deposits, where the flow of water was greater than at most of the mines.

None of the vertical shafts was equipped with ladders for the entry and exit of the men. The miners were let down into the mines in the round tubs used for hoisting the ore to the surface. At only a few of the mines were there separate shafts and hoists for the men, these also being used for lowering and hoisting powder, drill bits, or other supplies needed during the day. The average distance between shafts was a little less than 300 feet.

Generated on 2024-09-28 13:59 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

The cost of sinking shafts in the district varies according to the nature of the ground, but an average cost is about $10 to $12 per foot to a depth of 200 feet. This cost applies, of course, to the single-compartment shafts only.

DRILLING.

In all methods of drilling certain questions arise that have to be taken into consideration before choosing the kind of drill or the method of drilling to be used. First of all must be considered the nature of the rock in which the ore bodies are found—whether it is hard or soft, brittle or tough. The form of deposition in which it is found—whether in massive or stratified layers—is also important. Some rocks are creviced and in others there are large cavities, which oblige the drill man to place the holes so that the effect of the powder in breaking ground will be greatest. The chief object is to excavate the greatest amount of ore possible at the lowest cost, and in order to accomplish this end the above conditions must be considered before any estimates can be given. The lack of such consideration is one of the main reasons why some of the mine foremen can not understand why their men are unable to get out as much ore or obtain the same amount of footage in drilling as others do at other mines.

In the Joplin district most of the drilling is done with percussion drills driven by compressed air. When a drill hole runs into a cavity, loading is generally impracticable, so that a new hole has to be drilled. The rock is usually hard and brittle and rapidly wears the bits of the drills, but on account of its brittleness it is more easily broken by the concussion of the drill. The main rock of some of the ore bodies in this district is tough, and footage in drilling is consequently not as great, whereas in other deposits it is soft, so that less drilling is necessary to obtain the same tonnage. In a few of the "soft-ground" mines there are side drifts, in which the men drill holes 2 or 3 feet long with drill rods and hammer, using also picks for breaking down the face of the ore.

The length of the holes drilled varies according to the height of the face and the method of mining used. Where the face of the ore is about 10 feet high with no stoping and the rock is fairly hard, the holes are drilled 8 to 10 feet, the upper ones being about 1 foot shorter than those near the foot of the face. With underhand stoping, the holes for advancing the heading are usually 6 to 7 feet long, whereas those drilled in the underlying bench, which is called the "stope," are sometimes 16 to 17 feet long, the usual length being 12 to 14 feet. Compared with the general practice in metal mines these lengths are unusual. The average diameter of the holes is 3 to 3¼ inches at the mouth and 1½ to 1¾ inches at the back.

Generated on 2024-09-28 13:55 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

There are several makes of machine drills used in the district. The tendency at present seems to be toward the use of drills somewhat larger in size than have been used in the past because a greater efficiency is obtained from them. As has already been mentioned the drills are operated by compressed air, the author noticing only one mine where steam was in use. The compressors are usually rated roughly as furnishing a sufficient quantity of air to operate a certain number of rock drills, also in terms of cubic feet of air compressed per minute to a given pressure. The calculated average capacity of the compressors used in the district[a] is a little over 900 cubic feet per minute with a pressure of 90 to 100 pounds. This pressure, however, when it reaches the rock drills is considerably reduced, owing to losses in transmission or leakage in the pipes. At some of the mines the air compressors are overtaxed in order that as many drills as possible may be operated.

BLASTING.

The blasting of holes, as well as their drilling, must be given careful attention in order that the greatest efficiency in excavating ore may be obtained. As in the case of drilling, the miner or machine man should be familiar with the nature of the rock that is to be blasted and should arrange his drill holes to the best advantage to obtain the best results. He should ascertain the direction of the line of least resistance and drill the holes that are to be loaded with explosive in a direction nearly at right angles to that line or in a direction that will break the portion of the face that he wishes to remove. The arrangement of the drill holes should be such as to permit the easy handling of the rock drills and also to minimize the necessary number of holes and quantity of explosive. Two other important factors that should be considered in breaking ground are irregularities of the face and the form in which the face will be left after the shot. The importance of the latter detail relates not only to breaking a large quantity of rock, but also to the safety of the miners working in the drifts after each blasting, because of the danger from loose or partly supported pieces at the face.

Most of the machine men and ground bosses in the mines visited realized the importance of all of the above-mentioned considerations in the drilling and breaking of ground. The feature, however, that seemed to be most noticeable, especially where there was no stoping, was the lack of foresight as to the manner in which the face would be left after the blasting. In breaking the ground there seemed to be no definite system with reference to subsequent drilling and blasting so as to leave the face more accessible for the next round of shots. This observation does not apply to all of the men, but it is a detail that

[a] Based on the mines examined.

Generated on 2024-09-28 13:54 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

should be watched more carefully, and in many mines it would help to reduce the cost of explosives.

Next to labor explosives is the most important item in the cost of "hard-ground" mining in the district, and consequently their use to the best advantage always needs to be considered. High explosives are almost universally used throughout the district. The usual quantity of explosive kept on hand at the mines is 15 to 30 boxes of 50 pounds each, the miners taking underground each day enough for the day's shooting. It is stored on the surface in a magazine about 300 feet from the mill or other building. The magazines as a rule are constructed of wood with double walls, the intervening space between the walls being filled with tailings or manure. They are used not only for the storage of explosives, but also as thawing houses, and are equipped with steam pipes to keep dynamite at a temperature above its freezing point.

The explosive comes to the mines in 50-pound wooden boxes containing 80 to 140 cartridges, depending upon the kind of explosive used. It is lowered into the mines in the tubs used for hoisting the ore, usually three boxes being lowered at a time. At some of the mines there are single-walled magazines underground, but these are used only for the storage of fuses and caps or for the day's supply of powder.

The cartridges are prepared and loaded into the holes for the final blasting by a powder man, who has a helper in mines where a large number of holes are loaded each day; the chambering or "squibbing" of the holes is done by the machine men during the day. At nearly all the mines "squibbing" is done except in the "soft-ground" deposits. The average number of cartridges or sticks of explosive used in each hole varies according to the length of the hole and the strength of the explosive employed. The strength of the explosive that seems to be used most in this region is equivalent to that of a 40 per cent "straight" nitroglycerin dynamite. In wet holes gelatine dynamite is used; for dry holes an ammonia dynamite is coming into use more and more, and with good results.

After the powder men have prepared the cartridges by slicing the side of each "stick," and after the hole to be loaded has been blown out with compressed air, the holes are loaded and primed. This is done by shoving in the "sticks" of powder one at a time with a pointed wooden bar. When a sufficient number have been loaded into the hole, the primer or cartridge containing the cap (detonator) and fuse is inserted. Three different ways in which the cap is inserted in the primer are shown in figure 3.[a]

[a] The Bureau of Mines recommends that the detonator be inserted in the end of the primer as in the bottom diagram in figure 3. To lace fuse through a cartridge of high explosive is dangerous because the explosive, ignited by side spitting of the fuse, may burn before exploding, with the result that poisonous gases will be produced. Detonator should not be inserted so far in a cartridge as that indicated in figure 3.

Generated on 2024-09-28 13:56 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

From this point on the method of blasting varies somewhat in the different mines. The primer is usually followed by one or two "sticks" of powder which are tamped with a wooden bar. This course is supposed to give the best results and to do away with the necessity of stemming. Where this is not done, some men stem the loads with paper and dirt, whereas others do no tamping at all. At a few mines tamping bags or sticks of stemming material are used. The men as a rule do not care to use specially prepared stemming material, claiming that its use produces no better results and involves loss of time. This opinion seems somewhat unreasonable, for not enough experimenting with explosives has been done in this district to establish it.

In general the drilling and the "squibbing" of holes are done one day, the loading and blasting being done the day following. The machine men light their own shots after all other men are out of the mine. The time of shooting is about 4 p. m. or between 12 and 1 o'clock at night if there are two underground shifts. The amount of powder used per ton of ore broken varies according to the nature of the ground.

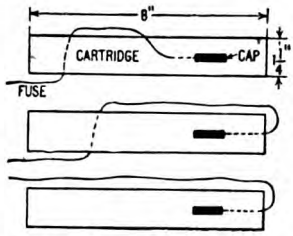

FIGURE 3.—Three ways of inserting detonator in primer.

SHOVELING.

After the ground has been broken, the dirt is shoveled into cylindrical tubs locally known as "cans." Three different sizes are in use, the most popular being 28 by 30 inches and 30 by 30 inches; at a few of the mines tubs 30 by 32 inches are used. The 30 by 30 inch tubs seem to be the most suitable, for the broken ore is easily shoveled into them and they are easy to handle and not likely to topple over when hauled to the shaft. For the convenience of the men known as shovelers, a small floor of wooden planks is laid so as to provide a smooth surface from which to work. The tubs are placed on small trucks and the tracks laid as far as the planks so that the broken ore can be shoveled directly into the tubs resting on the trucks. Bowlders are lifted into the tubs, being broken with sledge hammers when too large for one man to handle.

The shovelers are good workmen as a rule, as shown by the high average of 19.7 tons shoveled per man per shift. They are usually paid by the tub (4 to 8 cents per tub), so that they endeavor to get out as much as possible per shift. This system is good for obtaining tonnage, but it has one bad drawback in that the shovelers are oftentimes slack in filling the tubs in their effort to get a great number of tubs to their credit. This tendency can be overcome only by careful watching, and at many of the mines the ground bosses exercise proper supervision over this matter. As a result of the tendency of the men to shovel scant tubfuls and of the fact that

Generated on 2024-09-28 13:53 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

the filled tubs are rarely weighed, the tonnage of ore hoisted is often estimated considerably higher than it really is. Tubs 28 by 30 inches are said to hold an average of 850 to 900 pounds of ore and those 30 by 30 inches 1,000 to 1,100 pounds, whereas 800 to 850 pounds and 900 to 950 pounds would be nearer the average weights of ore hoisted per tub.

Instead of tubs, cars are used underground at some of the mines. The capacity of the cars is greater, but they are usually much heavier in weight and consequently require a better track bed and heavier rails, making the initial cost higher.

HAULAGE.

The underground systems of haulage are well adapted to the conditions in this district. Although two or three methods of hauling the ore from the working face to the shaft are in use, the most common method is for each shoveler, after having filled his tub, to push it on a small truck to the shaft or to a certain point or lay-by, whence it is taken to the shaft by special men known as "tub runners" or "mules," the shoveler returning to his place of work with an empty tub. When the face of the ore is only 100 to 200 feet distant from the shaft, this method works to great advantage, but when the distance is over 400 feet mules (the beasts) are used, thus effecting a great saving of time and cost in that instead of one tub at a time being pushed to the shaft, a train of five or six tubs can be hauled by one mule. Two of the mines visited use underground a system of rope haulage driven by electric motors. At one of the larger mines of the district, where cars instead of tubs are used, a gasoline locomotive has been installed for hauling the cars to the shaft, thus doing away with several mules.

TIMBERING.

The timber in the mines is not intended to resist the great pressure of the overlying rock, but is used mainly to support and keep in place any loose rock. Its cracking or gradual breaking also gives warning to the workmen enabling them to use additional timbering or to escape before a fall occurs. In the best practice the timbers are so placed that the pressure is evenly distributed and they are in the direction of the load that they are to resist. The joints are made so that the pressure, up to the crushing strength of the timber, tends to hold the structure together rather than to weaken it.

In this district, timbering is rarely used in either the "sheet-ground" or the "hard-ground" mines, for the roofs generally are good, and the pillars are left at intervals of 25 to 50 feet, according to the nature of the ground and the height of the roof above the floor. In nearly all of the "soft-ground" mines, however, timbering is employed to a great extent. As the drifts are advanced drift sets,

Generated on 2024-09-28 14:06 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

each composed of two vertical posts and a horizontal timber or cap, are put in and are connected by timbers of smaller diameter, laid horizontally next to the roof in the direction of the drift. Where the roof is rough or contains loose material, a filling of small sticks of wood and waste rock is placed across these horizontal timbers. The side posts are usually 10 to 12 inches in diameter and 6 to 8 feet long. Where the drifts are high, square-set timbering is generally used. Forepoling is used in the drifts of some of the mines.

In the "hard-ground" and the "sheet-ground" mines posts to support slabs are placed wherever necessary and in some of the mines cribs filled with waste rock are used as pillars to support the roof.

Oak timbers are most commonly used and are usually cut locally or shipped in from some near-by place.

PUMPING.

The amount of water to be handled at these mines varies considerably. The levels of the "sheet-ground" mines are fairly dry throughout the district, whereas those of many of the soft-ground deposits are wet. Although the levels may be dry, the drifts along the tracks often have water 1 to 2 inches deep, with no especial drainage system except whatever paths the water can find to the sump from which it is finally pumped. Usually the water gets into the mine workings from some watery stratum or as a result of intermittent seepage, and unless the workings possess a natural drainage, the seepage accumulates and has to be pumped out. The latter condition is general in the district for the mines are shallow. At a few mines underground water passages have been met which necessitate heavy pumping. At nearly all the mines the water is drained into a main sump at the bottom of some shaft by means of ditches cut along the sides of the drifts. Many of the shafts are wet as a result of seepage. In a few mines a second sump is dug, the water being pumped from this sump into the main sump and then to the surface.

Because of the shallow depth of the mines, it is often difficult to handle the water incident to a heavy rainfall, so that when the pumping equipment is insufficient, the mines must be shut down until most of the water has been pumped out.

SURFACE WORK.

HOISTING.

The system of hoisting most common in this district is that of direct winding without attempt to balance the load. The bucket and its load are hoisted by an engine and lowered by gravity. The engine is coupled or geared direct to the shaft of the drum, which is usually provided with friction devices or positive clutches and brakes.

Generated on 2024-09-28 13:57 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

Hoisting by balance, as employed at a few of the mines, is usually effected by the use of double cylindrical drums.

At the foot of the shafts wooden platforms are built on which to place the buckets or tubs just before they are hoisted to the surface. At four of the mines visited cages or skips were in use. These were self-dumping and had shaft guides, but only in two of the shafts were they equipped with safety catches. Their capacity, 1 to 2 tons per load, is considerably greater than that of tubs.

The hoisting distance is 150 to 250 feet, and where the loaded tubs at the foot of the shaft are always ready the round trip can readily be made in 35 to 45 seconds. When the tub has been hoisted to the top of the derrick the hoist man covers the opening with a lid to prevent any loose material from falling back down the shaft and then dumps the load on a chute leading directly to the grizzly over the hopper. The headframe timbers extend above the breaker platform; they are of the four-post type and are locally known as "samson" posts. They are about 30 feet high and are properly braced. The derricks are of wood. Few are housed because of the danger from fire.

The power most commonly used for hoisting is steam, but a few of the mines are equipped with electric hoists, which have given satisfactory results.

Signaling for hoisting and lowering both men and ore is done by a pull bell, although a few mines have installed electric bells. In hoisting the men, as a rule, not more than four at a time are permitted to ride in the buckets. Each man stands on one leg inside the tub, his other leg being outside. This way of riding is not without risk, for there is always danger of injuring the kneecap or leg if the tub should bump against the sides of the shaft while being hoisted or lowered.

SURFACE TRAMMING AND STORAGE OF ORE.

At these mines each shaft is built adjacent to a storage bin or hopper, and with few exceptions the main hopper is connected directly to the mill. Where hoisting is done in more than one shaft the ore is trammed from the foot of each hopper up an inclined tramway directly into the mill hopper or from one bin to another until it reaches the mill hopper.

The ore, after having been hoisted from the mine and passed through the grizzly into the bin below, is drawn through the loading gate at the bottom of the bin into a wide-gage tramcar. These cars are self-dumping and have a capacity of 1 to 2 tons. They are hauled up the incline by a hoisting engine and return by gravity to the foot of the tramway to be reloaded from the hopper. At a few small mines, where the shaft is close to the mill hopper, there is an

Generated on 2024-09-28 13:53 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

overhead tramway and the filled buckets are unhooked from the hoisting cable, dropped on trucks similar to those used underground, pushed to the hopper, and dumped. This method of tramming consumes too much time. The hoppers to which the ore is hoisted and trammed are built of wood, their capacities varying from 100 to 300 tons. Those of the greater capacity are the mill hoppers, which are often built large in order to hold an amount of ore sufficient to supply the mill during an extra shift. This extra capacity is rather common at the larger mines throughout the district.

POWER.

Natural gas, electricity, coal, and oil are all used in this district as fuel for generating power, one of the main objects of the operators being to obtain the cheapest power possible. Of the mines visited, gas was used exclusively for fuel throughout the year at 25, electricity at 7, and coal at 2. Gas was used in combination with other fuels at 45 mines, electricity at 13 (not including mines equipped with electricity for lighting purposes only), coal at 16, and oil at 5. Coal and oil were generally used only during the winter months when the gas pressure was low.

The mines that use steam power are equipped with two or three boilers of 100 to 150 horsepower each, which supply the steam to the compressors, hoisting engines, and engines for running the mills. Instead of steam engines many gas engines are used for driving the mills, natural gas for gas engines being furnished at 25 cents per 1,000 feet. Natural gas for fuel under boilers costs 12½ cents per 1,000 feet and is supplied from Oklahoma and Kansas. Electricity is gradually coming into use and some of the new properties are being equipped entirely with electric power.

ORE-DRESSING METHODS.

For convenience in discussing the methods of ore dressing or concentration used in this district, a diagram (fig. 4), with explanatory legend, is presented. This shows in approximate order the different steps through which the ore passes and the approximate arrangement of the mill. The diagram is a fair average of the mills in the district, the arrangement of apparatus and their measurements as given in the legend being based on the concentrating equipment at 61 mills. The flow sheet of one of the larger mills in the district, together with supplemental explanatory legend, is given in figure 5. Reference to the italic figures in the diagram showing the mill arrangement (fig. 4) is made in the subsequent discussion.

Generated on 2024-09-28 13:56 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

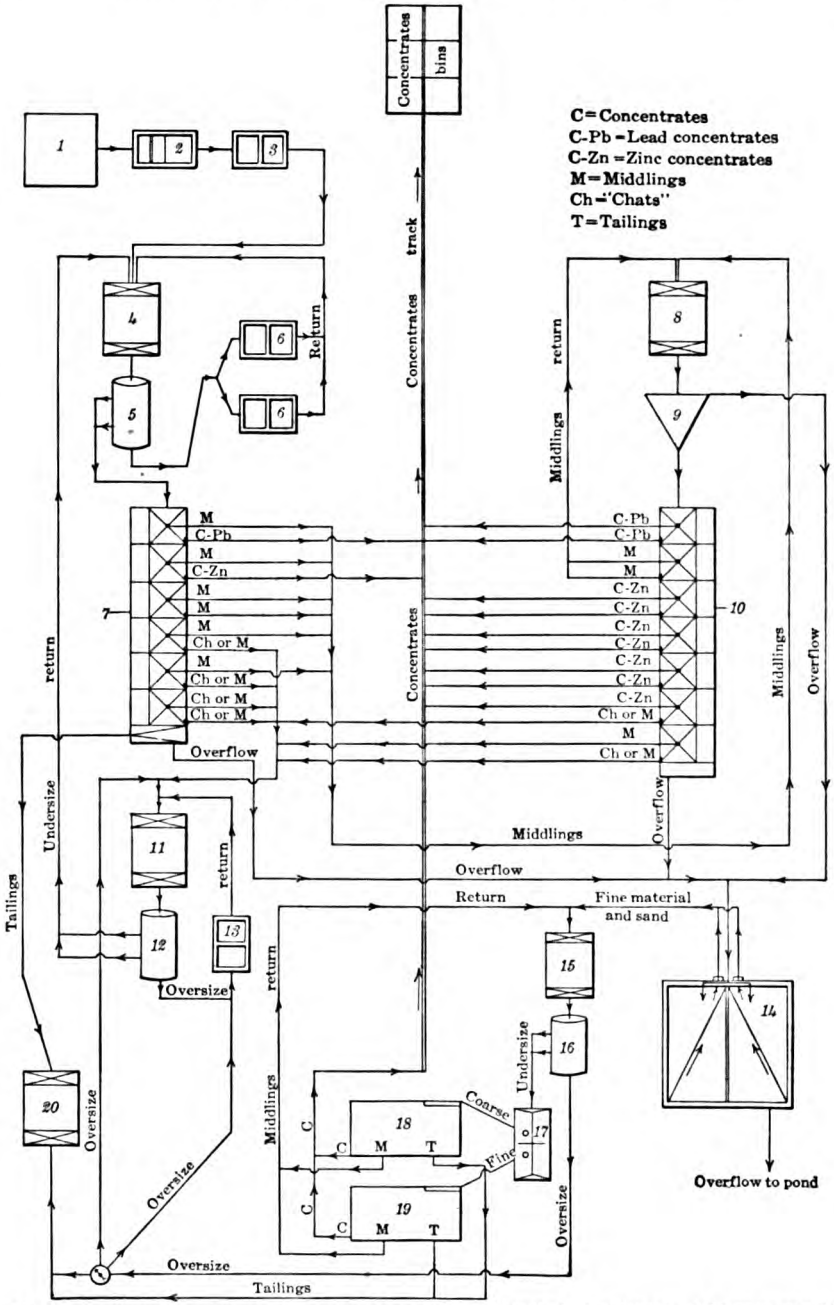

FIGURE 4.—Arrangement of an average zinc concentrating mill in the Joplin district. *1*, Hopper, capacity 225 tons; *2*, crusher, 16-inch Blake type; *3*, rolls, 36-inch; *4*, elevator, 18 by 7 inch buckets; *5*, trommel, 48 by 96 inches, with ½-inch round perforations; *6*, rolls, 36-inch; *7*, rougher jig, six 36 by 48 inch cells; *8*, elevator (middlings), 12 by 7 inch buckets; *9*, dewatering box (fine sand and overflow); *10*, cleaner jig, seven 30 by 36 inch cells; *11*, elevator ("chats" and middlings), 10 by 7 inch buckets; *12*, trommel, 36 by 48 inches with ⅛-inch perforations; *13*, rolls (for "chats" and middlings), one set, 24-inch; *14*, settling tanks, two, each 10 by 20 feet, wood; *15*, elevator (sand and fine material), 10 by 7 inch buckets; *16*, trommel, 36 by 60 inches, with 1½-mm. perforations; *17*, settling box, two compartments (coarse and fine material); *18*, table (coarse material); *19*, table (fine material); *20*, elevator (tailings).

Generated on 2024-09-28 13:56 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

CONCENTRATING MILLS.

The topography of the Joplin district and of the surrounding country is generally flat, so that the concentrating plants, or mills are of the level-site type, elevators being employed to raise and advance ore from one treatment to the next.

Although the mills are in general of similar design they differ in many details, more so than is apparent to the casual observer. No large mills have been erected, for, as a rule, no attempt is made to block out a supply of ore in the mines sufficient to keep a mill running for a considerable length of time after it has been erected. Most of the mills are constructed of wood, although a wood frame covered with galvanized iron, with pillars or posts of concrete for the foundation, is used in a few mills. On the whole the equipment is poorly housed and during very cold weather milling operations often have to be suspended because water freezes in the pipes and tanks, and ice forms on the elevator pulleys, causing the belts to slip off. As a rule, however, the cold weather is of short duration, so that most of the mining companies do not consider it worth while to employ better housing. The capacity of the average mill is about 150 to 200 tons per 10-hour shift, although some have a much greater capacity. The course of the ore treatment, as outlined by the flow sheet, is usually simple.

As has already been mentioned, mills are nearly always built adjacent to a shaft, usually the main shaft, so that the ore can be hoisted directly into the mill hopper. This construction simplifies the handling of the ore, thus reducing costs and lessening the quantity of fine material caused by abrasion.

The cost of constructing the mills, including equipment, is from $10,000 to $30,000, depending on the capacity and the kind of machinery and concentrating equipment installed, although the cost of a few of the larger and more recently built mills is higher.

COARSE CONCENTRATION.

SORTING.

When the ore has been hoisted and dumped on the chute and has fallen to the grizzly bars over the bin or hopper the largest pieces are caught on the bars. The bars are usually heavy rails placed 4 to 5 inches apart, and bowlders of greater diameter than that are handled by one or two men, known as cull men, who stand at the bars. After each tub of ore has been dumped, the cull men break with hammers the bowlders that are sufficiently mineralized; the bowlders that are practically barren they place in a car at the side, run the loaded car out to the end of a tramway, and dump its contents on the waste pile. The amount of rock sorted out in this manner varies from 1 to

Generated on 2024-09-28 13:55 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

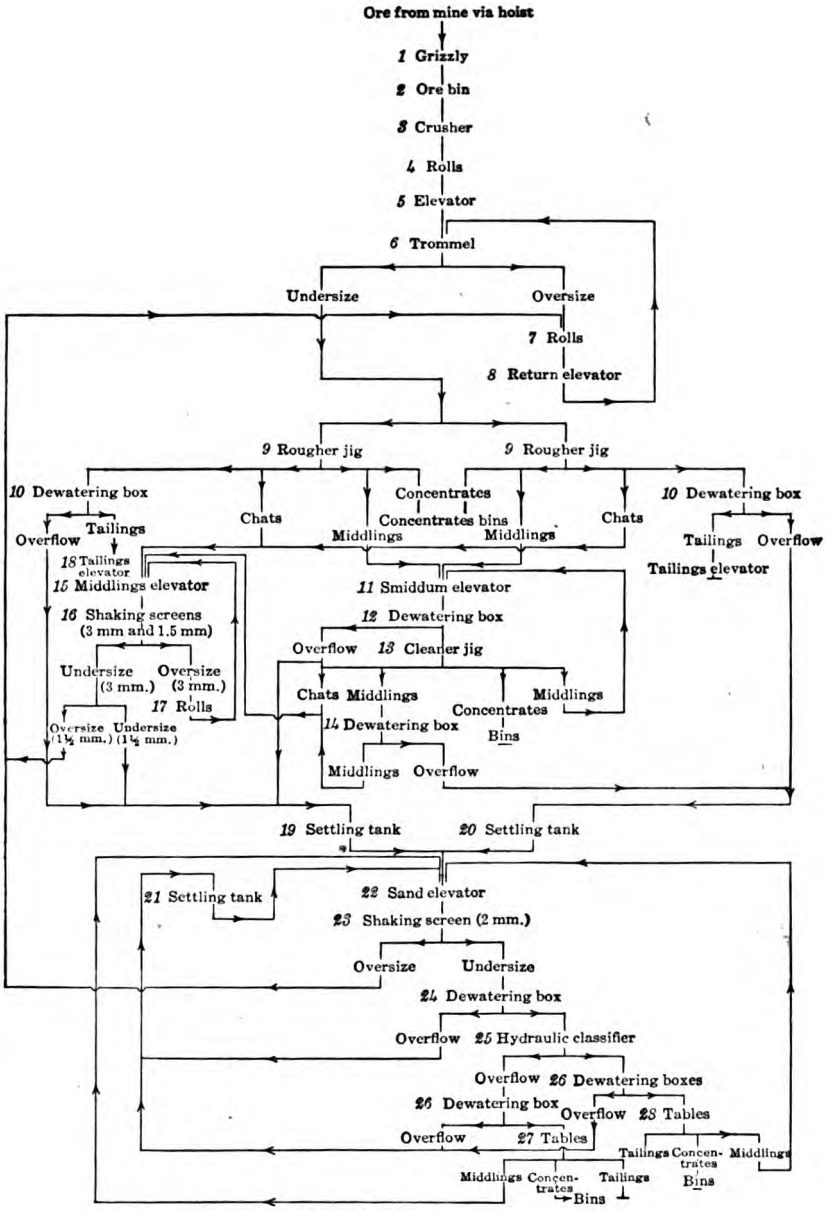

FIGURE 5.—(For description see page 25.)

Generated on 2024-09-28 13:56 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

FIGURE 5.—Flow sheet of large zinc mill in the Joplin district. *1*, Grizzly, bars 5 inches apart; *2*, ore bin, 25 by 30 by 25 feet, capacity 500 tons; *3*, crusher, 18 by 8 inch opening, 385 revolutions per minute, driven pulley 44 by 12 by $3\frac{7}{16}$ inches, drive pulley 30 by 12 by $3\frac{7}{16}$ inches, 6-ply belt 61 feet 8 inches by 12 inches; *4*, rolls, 1 set, 14 by 36 inches, 30 revolutions per minute, driven pulley 44 by 12 by $3\frac{7}{16}$ inches, drive pulley 30 by 12 by $3\frac{7}{16}$ inches, geared ratio 19 to 86, spur 56 by 6 by $5\frac{1}{16}$ inches, 86 teeth, pinion $12\frac{1}{2}$ by 6 by $3\frac{7}{16}$ inches, 19 teeth, 6-ply belt 48 feet 2 inches by 12 inches; *5*, elevator, 20 inches wide by 23 feet high, 7 by 10 inch buckets spaced 10 inches apart on alternate sides of 10-ply belt, 53 feet 6 inches by 20 inches, speed 240 feet per minute; *6*, trommel, 2 sections, each 48 be 156 inches, $\frac{7}{16}$-inch round perforations, slope $1\frac{3}{8}$ inches per foot, speed 21 revolutions per minute, shaft 12 feet 6 inches by $3\frac{1}{16}$ inches, bevel gears; *7*, rolls, 2 sets, 14 by 36 inches, speed 30 revolutions per minute, 6-ply drive belt 54 feet by 12 inches; *8*, return elevator, 20 inches by 28 feet high, 7 by 10 inch buckets spaced 10 inches apart on alternate sides of 8-ply belt 64 feet by 20 inches, speed 300 feet per minute; *9*, rougher jigs, two, Cooley type, each having 5 cells 34 by 48 inches, 118 revolutions per minute, grates of perforated sheet steel, perforations $\frac{1}{8}$ by $\frac{9}{16}$ inch, sheets $33\frac{1}{2}$ by $47\frac{1}{2}$ inches, 6-ply drive belt 29 feet by 12 inches; *10*, dewatering boxes, two, for water and fine sand overflow; *11*, "smittem" elevator, 18 inches by 19 feet high, buckets 7 by 10 inches, spaced 9 inches apart, 8-ply cup belt 44 feet 6 inches by 18 inches, speed 380 feet per minute, top pulley 24 by 18 by $2\frac{1}{16}$ inches, shaft 6 feet 2 inches long; bottom pulley 24 by 18 by $2\frac{1}{16}$ inches, shaft 5 feet long, gear spur wheel 24 by $3\frac{1}{2}$ by $2\frac{1}{16}$ inches, pinion 9 by $3\frac{1}{2}$ by $2\frac{1}{16}$ inches, 6-ply drive belt 19 feet 11 inches by 8 inches; *12*, dewatering box for water and fine-sand overflow; *13*, cleaner jig, Cooley type, 166 revolutions per minute, 7 cells each 30 by 42 inches, grates of perforated sheet steel $29\frac{1}{2}$ by $41\frac{1}{2}$ inches, perforations $\frac{1}{8}$ by $\frac{9}{16}$ inch, 6-ply drive belt 29 feet 5 inches by 12 inches; *14*, dewatering box for water and fine-sand overflow; *15*, middlings elevator, 16 inches wide by 20 feet high, buckets 7 by 16 inches, spaced 10 inches apart, 6-ply cup belt 45 feet 6 inches by 16 inches, speed 310 feet per minute, top pulley 24 by 16 by $2\frac{1}{16}$ inches, shaft 6 feet 4 inches long, bottom pulley 24 by 16 by $2\frac{1}{16}$ inches, shaft 5 feet long, 6-ply drive belt 28 feet 2 inches by 8 inches; *16*, shaking screens, two, 3-mm. and 2-mm. perforations; *17*, middlings rolls, one set, 19-inch, 13 by 19 inches, speed 28 revolutions per minute, gear spur-wheel 36 by 4 by $3\frac{1}{16}$ inches, 76 teeth, pinion 9 by 4 by $2\frac{7}{16}$ inches, 19 teeth; *18*, tailings elevator, 24 inches wide by 73 feet high, buckets 7 by 12 inches spaced 10 inches apart on alternate sides of belt, 8-ply cup belt 153 feet by 24 inches, speed 385 feet per minute, top pulley 30 by 24 by $3\frac{1}{16}$ inches, shaft 6 feet long, bottom pulley 24 by 24 by $3\frac{7}{16}$ inches, shaft 4 feet long, gear spur-wheel 36 by 4 by $3\frac{1}{16}$ inches, 76 teeth, pinion 9 by 4 by $2\frac{1}{16}$ inches, 19 teeth, 6-ply drive belt 107 feet 4 inches by 10 inches; *19*, settling tank, continuous draw-off, 6 by 11 by 4 feet deep in center; *20*, settling tank, continuous draw-off, 5 by 17 by 5 feet deep in center; *21*, settling tank, continuous draw-off, 6 by 11 by 4 feet deep in center; *22*, sand elevator, 14 inches by 36 feet high, buckets 7 by 14 inches, spaced 9 inches apart, 6-ply cup belt 79 feet by 14 inches, speed 345 feet per minute, top pulley 24 by 14 by $2\frac{1}{16}$ inches, shaft 4 feet 8 inches long, bottom pulley 24 by 14 by $2\frac{1}{16}$ inches, shaft 3 feet 6 inches long, 5-ply geared drive belt 26 feet 7 inches by 8 inches; *23*, shaking screen, 2-mm. perforations; *24*, dewatering box; *25*, hydraulic classifier, 3 compartments; *26*, dewatering boxes; *27*, tables, two, Wilfley, for fine material (overflow from classifier); *28*, tables, two Ford tables for coarse material and two Wilfley tables for intermediate material.

Generated on 2024-09-28 13:58 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

15 per cent of the total tonnage hoisted, the average being about 5 per cent, depending on the nature of the deposit. The average cost of handling the waste rock is about 2.1 cents per ton of ore hoisted.

Sorting other than that done by the cull men is rarely seen at the mines in the district. At one of the "sheet-ground" mines hand picking of the ore just before it enters the crusher is employed. The superintendent of this mine stated that the amount of waste rock so eliminated was such that the percentage of mineral content in the retained ore was raised and the capacity of the mill relative to the output of the mine was increased, the resultant saving warranting the extra sorting. Such additional sorting is to be adopted on a large scale at this mine, and other companies are also said to be contemplating its adoption.

CRUSHING.

The undersize material, 6 inches or less in diameter, from the ore dumped on the grizzly bars falls into the mill hopper (*1*, fig. 4). The capacity of these hoppers or receiving bins varies according to the tonnage handled, the average being 225 tons. From the hopper the ore is fed onto an incline leading to a crusher (*2*), that reduces the larger material to pieces about 1-inch size or less, depending upon the setting of the jaws of the crusher at their lower end. At a few of the mills, before entering the mouth of the crusher, the material first passes over a perforated screen having ¾-inch holes. A stream of water plays upon the material as it passes over the screen. The finer material is washed through, thus eliminating unnecessary crushing and increasing the capacity of the crusher relative to the tonnage treated. The undersize from the screen goes to a set of rolls or directly to the elevator, where it is raised to the trommel at the head of the rougher jig. The crushers are manufactured locally and are of the Blake type.

After the material has been crushed to 1-inch size or less it is fed directly into a set of rolls (*3*) just below the crusher and is largely reduced to ¾-inch size or less. As a rule, only one set of rolls is used, although two sets are not uncommon.

The ore next goes to an elevator (*4*), by which it is raised to a trommel (*5*). These elevators are of the usual gear-driven belt type, equipped with conveyer buckets spaced about 10 inches apart. The ore, as it reaches the top, falls into a receiver on the trommel (*5*). Where tonnage is large two trommels are used, or one with two sections of the same dimensions. As the ore falls on the revolving screen, the ½-inch undersize passes through directly to the rougher jig (*7*); the oversize goes to a set of return rolls (*6*), by which it is further crushed and is then carried by the elevator (*4*) back on to the trommel (*5*). In this manner all of the ore is reduced to ½-inch size before it finally reaches the rougher jig.

Generated on 2024-09-28 13:53 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

JIGGING.

The most common type of jig used in this district is known as the Cooley jig, similar in principle to the Harz jig. The jigs do very good work if the large quantity of material that passes over them be considered. They are of the fixed-sieve type, the water being forced up and down through the screens or grates by the action of a plunger. This plunger is placed in an adjacent compartment connected with the hutch below the grates and receives its motion from an eccentric revolving on a shaft which is connected to a battery of plungers.

The material, together with considerable water, comes on to the first cell of the rougher jig (*7*) from the revolving trommel (*5*). A bed 5 to 7 inches deep is formed, and as a result of the pulsating action of the plunger the lighter material, such as flint, limestone, dolomite, and calcite, settles on the surface, the heavier free grains of lead and zinc minerals working to the bottom. The downward suction stroke of the plunger causes the finer grains of mineral and sand to continue through the openings into the hutch. The strength of the suction stroke is increased because the gates of the hutch are left partly open. The accumulation in the hutch is known as "smittem," or middlings, and is re-treated on the cleaner jig (*10*). From the first cell, as well as from the second, are produced the first concentrates, which go direct to the concentrates bins. No finished concentrates are obtained from the rougher jigs in some of the mills. Between each cell there is a fall of a few inches, so that the material gradually advances from one to the next until it reaches the sixth and last. As the material advances its mineral content decreases until from the last cell the bulk of it flows off as tailings into the tailings elevator (*20*). The other products from the rougher jig are middlings and "chats," and they either go directly to the cleaner jig for re-treatment or are first sent to a set of rolls for finer crushing.

Middlings consist of particles of free ore mixed with particles of gangue or mixed with gangue particles containing mineral matter; "chats" consist of gangue particles containing ore that has not been separated by the first crushing. In regrinding, the middlings or the "chats" go first to an elevator (*11*), which raises the material to a revolving trommel (*12*) with $\frac{1}{8}$-inch perforations. The undersize from the trommel is returned to elevator *4;* the oversize goes to a set of rolls (*13*) to be recrushed. From the rolls it returns to elevator *11*, so that all of the middlings and the "chats" from the rougher jig that are recrushed return to the original lot of material being treated. This procedure is not followed in all the mills of the district, for when the quantity of "chats" to be handled after regrinding is large it goes to a smaller jig, where it is treated separately. This is by far the better method, for if the "chats" and the middlings are

Generated on 2024-09-28 13:57 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

returned to the original lot of material the quantity to be handled at the first concentration is increased, and the losses in the tailings from the rougher jig are liable to be considerably greater. This result is especially to be expected at mills where the capacity of the jigs is overtaxed. In order to obtain the desired capacity from the rougher jig, the tendency is to increase the flow of water, rushing the material through as quickly as possible. At several of the mills visited the tailing from the rougher jigs contained small particles of free ore, which the heavy flow of water had undoubtedly taken with it, the particles not having time to settle or work themselves into the lower part of the bed. The losses in "chats" are still greater, for the difference in specific gravity between chats and many somewhat larger gangue particles is slight and they are consequently difficult to separate by jigging.

At the end of the rougher jig there is a box into which the overflow and tailings fall. The box has at its lower end a spigot through which the tailings pass into elevator (*20*). The spigot serves to check the flow of material so as to cause the box to fill with water and overflow. The overflow, as it comes from the last cell of the jig, usually contains fine grains of free ore and is therefore sent to settling tanks (*14*) for further treatment. These tanks are usually placed just outside of the mill. They are discussed more fully in a subsequent section of this report.

The middlings from the rougher, from which the bulk of the concentrates is obtained, go to an elevator (*8*) which advances the material to a dewatering box (*9*). The material sinks to the bottom of this tank and flows to the cleaner at a uniform rate; the overflow goes to the settling tanks (*14*).

The process of jigging on the cleaner jig is the same as on the rougher jig. The compartments are usually smaller and the action of the plunger is considerably faster. The beds of ore on the cleaner jig are not as heavy and the quantity of water used is not as great. The enriched material from the rougher jig receives its last concentration on the cleaner. It reaches the first compartment where the product of lead concentrates is obtained from both the hutch and the bed of the first cell. If there is a reasonable proportion of lead mineral to that of zinc in the ore, the lead concentrates from this first compartment are usually very clean, averaging about 80 per cent lead content. From the second compartment a middling is produced consisting of a mixture of lead and zinc concentrates, which is returned to the return elevator (*8*) and re-treated. Zinc concentrates in final form are obtained from the other compartments of the cleaner jig except that the beds of the last two compartments and the hutch of the last compartment yield middlings or "chats" which are sent to

Generated on 2024-09-28 13:58 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

rolls for recrushing. The overflow water from the cleaner jig contains fine mineral matter and goes to the settling tanks (*14*).

In a few of the mills the finer middlings are sent to sand jigs instead of being returned to the jig from which they came.

CONCENTRATION OF FINE MATERIAL.

In the concentration of the fine material the operators of this district have, until the past few years, been rather behind in their methods, but are beginning to realize the importance of saving the finer mineral particles formerly wasted in the discarded sands. There are still several mills in the district that have no concentrating tables or sand jigs, no attempt being made to avoid the losses mentioned. The practice in treating the fine material is discussed below.

The fine material, consisting of sand and mineral matter that can pass a 2 to 3 mm. screen, flows into the settling tanks (*14*). The settling tanks are built of wood and are placed just outside of the mill. There is usually only one large tank, 20 by 20 feet, divided into two equal sections, so that while one is being emptied the other can be filled. The material enters the tank at the end nearest the mill. The sand and fine grains settle to the bottom and the overflow water discharges at the farther end. After the water in the tank has been drained a small stream of water is used to wash the material down the sloping floor of the settling tank through a gate or spigot into an elevator (*15*), which raises it to a point where it flows on to a trommel (*16*) with 1½ mm. to 2 mm. perforations. In place of trommels, shaking screens are used in some mills, but these have rarely given satisfactory results. The oversize from the screen or trommel (*16*) goes either to the tailings elevator (*20*) or to the "chats" or middlings rolls (*13*) for finer crushing; the undersize flows to a settling box (*17*) before going to the tables. At the mills where the oversize goes to the tailings elevator the operator or mill man clearly does not consider it to have any appreciable value. However, as in the case of tests 58 and 59 (p. 35), the oversize often contains zinc mineral in amounts worth saving. In the sizing test made from the general tailings (p. 35) nearly 19 per cent of the total loss in the tailings was in the sizes between 3 mm. and 1.5 mm.

As has been stated, the undersize from the revolving screen goes to a settling box or spitzkasten (*17*) before going to the tables. The procedure at this point, however, varies considerably in the different mills. At a few there are classifiers that give good results, but on the whole the classification of the finer material is rather poor, and at many mills there is practically none. Hydraulic classifiers of the Richards type were being used in a few mills, others being of local design. In general, however, the classification is done in

Generated on 2024-09-28 13:55 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

settling boxes, partitioned off into sections, so that as the material enters the coarse particles fall into the first section and the fine particles into the second, or if there are three sections the material is divided into coarse, medium, and fine particles. Where the small particles are to be classified without any upward hydraulic movement a clean separation is difficult. Instead of having one settling box divided into sections for the different tables there are often two or more boxes, one for each table, which are so arranged that the overflow from the first goes to a second, thence to a third, and so on, depending upon the number of tables used. This system gives a fairly good classification if properly arranged. Others have only one settling box, with no sections, at the bottom of which are spigots for the different tables, scarcely any classification being obtained.

TABLE WORK.

From the settling boxes or classifiers the material goes to the tables (*18* and *19*) for concentration. The concentrating tables used in this district are of various types, differing, however, principally as to table deck and stroke.

In the concentration of fine material on tables certain factors must be considered in order to obtain good results. One of the most important of these is a uniform supply of the material fed to the tables, a requisite not only in obtaining a good product of concentration but also in keeping the losses in the tailings reduced to a minimum. One of the main faults with the table concentration in the Joplin district is the lack of uniformity in the supply of feed to the tables as it passes from the settling tanks (*14*) through the elevator (*15*) and the trommel (*16*) to the settling box (*17*); another is the lack of attention given the settling-box spigots, which frequently become clogged.

When the quantity of material fed to a table is suddenly increased, a larger quantity moves forward, producing a poor line of separation and more middlings, and the bed increases in depth so that the finer grains of mineral flow off with the waste sand at the tailings end of the table. Tables require constant attention to insure the best results. If the quantity of material is increased or decreased, the line of separation between two minerals of different specific gravities will move up or down accordingly at the end of the table. This is especially true of tables on the decks of which the riffles extend the full length, and when this line of separation is not watched carefully a poor product is sure to result.

Many of the tables, after being placed in the mills, are tested for a speed and stroke that will yield a good product. After a short time, however, little attention is paid to these factors, and, with few exceptions, no periodic test is made of the efficiency of the tables by accu-

Generated on 2024-09-28 13:56 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

rately sampling the feed and tailings from them. The stroke and the speed of the tables were determined in nearly all the mills visited, and it was clear that many of the poor results obtained could have been improved by simply increasing the speed without altering the stroke, or by changing the stroke, the present speed being maintained. To determine the speed and stroke best adapted for a given table, experiments must be made until the most efficient combination is reached. Another important consideration is the quantity of material that each table is to handle.

From the various tests made of the tables the best combinations for the material treated in this district were found to be as follows: For coarse material a speed of 220 to 240 strokes per minute, with a stroke of three-fourths to seven-eighths of an inch in length; for medium-sized material a speed of 240 to 260 strokes per minute, with a stroke of five-eighths to three-fourths of an inch in length; and for fine material a speed of 250 to 280 strokes per minute, with a stroke of one-half to five-eighths of an inch in length. Where the material was not classified at all and especially where only one table was used, the best combination of speed and stroke seemed to be a speed of 230 to 250 strokes per minute, with a three-fourths to seven-eighths inch stroke, depending on the quantity of material to be treated. In cases where the stroke was too short or the speed too slow, it was noticed that beneath the bed on the table near the feed end a concentrated product of fine grains of blende gathered in the riffles and was gradually swept down with the tailings because of the insufficient length of stroke or the insufficient speed. In test 51 (see p. 35) it will be noticed that the assays from the tables were high, especially those of the fine material. The length of stroke for the three tables was three-fourths of an inch, with speeds of 254 strokes per minute for the coarse, 248 strokes per minute for the intermediate, and 230 strokes per minute for the fine materials, these speeds being the reverse of what they should have been. The high results of the assay are also partly due to poor classification, for the first table was being crowded, whereas the third and largest table of the three was treating scarcely any material at all except slimes, so that the losses were not as great as the figures seem to show, the greatest loss being from the first table, which was treating coarse material.

As the speed of a table depends on the stroke, or vice versa, so the effect of tilting the table at right angles to its length depends on the quantity of water used to wash the material. Here, again, the slope or incline of the table and the quantity of water used should depend on the quantity of material treated on each table. The quantity giving the best results should be determined experimentally. At a few of the mills the quantity of water used at the feed end of the table was much too great, so that the fine grains of mineral were being washed

Generated on 2024-09-28 13:55 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

off with the tailings, as they did not have sufficient time to settle or work forward into the bed. A better separation can frequently be made by simply changing the slope of the table or the quantity of water used.

Another important detail in the treatment of fine material is the re-treatment of middlings from the tables. In nearly all the Joplin-district mills, with a few exceptions, the middlings from the tables are returned to the sand elevator and back to the same tables from which they came. This course not only increases the amount of material handled by each table, but increases the richness of the material fed to the tables, and consequently results in greater losses in the tailings. Whenever possible, if the quantity of fine material is sufficient to warrant it, the middlings from tables should be re-treated on separate tables rather than returned to the tables from which they originally came. In most mills where tables are used at least one extra table could be installed with advantage to treat the middlings separately, and without doubt would result in a higher recovery. The principle here is practically the same as in the case of returning the middlings and "chats" over the same rougher jig in the concentration of coarse material.

If the above-mentioned details of treating the finer material on tables were considered more carefully by the men in charge, much better results and a higher recovery could be obtained, for any standard table will do good work with proper care and attention. Although many of the tables are doing excellent work, it must be said that "sludge men," or those in charge of the tables, who really understand the important factors in obtaining good results from tables, are scarce in the Joplin district. It is expected, however, that as more tables are placed in the mills the work will be improved and the losses decreased.

HANDLING AND CHARACTER OF THE CONCENTRATES.

The finished products from the jigs are shoveled into a car, which is pushed on a track running the length of the mill, and to a trestle, where the concentrates are dumped into the bins. These cars are weighed, and are usually loaded with about 1,000 pounds of concentrates. The bins are on each side of the trestle and are used to store the products until they are shoveled into wagons and hauled away for shipment.

The zinc concentrates throughout the district assay 54 to 63 per cent zinc content. The average percentage of zinc in the concentrates from the mills visited was 59.2 per cent, although the average for the district is somewhat lower. The quantities of impurities vary. Five-tenths of 1 per cent to 4 per cent of iron sulphide and 0.5 to 1 per cent of lead is present, and some of the concentrates contain small quantities of lime.

Generated on 2024-09-28 13:53 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

SOURCE AND QUANTITY OF WASTE.

As the cost of timbering in the district is high, pillars are left in the mines to support the roof. From 10 to 25 per cent of the deposit is so left, the average being about 15 per cent, and although more or less trimming of the pillars is done after the mines have been practically worked out the ore left as pillars is often a total loss. Some of the mines have, just below the deposit being worked, a deposit too thin to be mined with profit under existing conditions.

The greatest waste, however, is in the milling or concentration of the ores. The losses in concentrating the lead ores is not as great as in the case of the zinc ores. The losses in milling zinc ores in this district have been extremely large, and have been due to the wasteful methods used. The main object of the operators is to treat as many tons of ore as possible in the shortest time and at the lowest cost. They are obliged to rush the ore through the mill as fast as possible in order to obtain enough concentrates to meet expenses and high royalties and still make a profit, a demand that has been met in a great many cases by a heavy loss of mineral.

The average recovery of blende from the ore in milling is about 60 to 65 per cent. The average loss, therefore, in the concentration of the zinc ores is 35 to 40 per cent. When the heavy losses in the smelting of zinc ores are included, the total loss in the production of zinc, from the ore in the mine up to the commercial product, reaches nearly 50 per cent. Very few operators in the district really know the percentage of recovery from their mills, because they do not make systematic mill tests. It is hoped, however, that they will make more of an effort to find out where the losses are, although in many cases it would be difficult, and realize the importance of a greater saving in the milling, so that the average percentage of recovery will gradually be raised several points. There are many tailing piles in the district that still contain from 2 to 3 per cent blende, and although some are being re-treated with a profit, much of the waste represented could have been saved in many cases by better methods in the original treatment of the ores.

MILL TESTS.

In connection with investigation of the coarse and the fine concentration in various mills in the district, certain tests were made, the results being tabulated below. Samples were taken every 15 to 20 minutes, the sampling being continued 5 to 7 hours, thus insuring a representative sample of the day's run. Each pile of samples was quartered down to a quantity of 15 to 20 pounds, the material being below one-half inch in size, careful mixing being done after each quartering. The accuracy of this method of taking a final sample

Generated on 2024-09-28 13:53 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

from the main bulk was tested by taking two parts from the same final sample. The samples were sent to two different assayers, who reported practically the same results. Accurate sampling in the mills, however, is difficult, and the making of a true mill test in any of the mills is almost impossible, so that the tests were confined to the separate steps in the milling. They gave a fair idea of where the losses were. The results and the character of the tests are presented in the following tabulations:

Results of tests of ore from mines other than "sheet-ground" mines.

Test No.—	Zinc in material fed to rougher jig.	Zinc in tailing from rougher jig.	Test No.—	Zinc in material fed to rougher jig.	Zinc in tailing from rougher jig.
	Per cent.	*Per cent.*		*Per cent.*	*Per cent.*
1	5.00	1.52	8	3.37	1.67
2	4.40	1.85	9	1.70	.65
3	3.40	1.05	10	5.25	2.00
4	4.35	2.02	11	2.37	.90
5	4.05	1.55	12	2.95	.87
6	1.32	.72			
7	4.70	1.55	Average	3.57	1.36

Results of tests of ore from "sheet-ground" mines.

Test No.—	Zinc in material fed to rougher jig.	Zinc in tailing from rougher jig.	Test No.—	Zinc in material fed to rougher jig.	Zinc in tailing from rougher jig.
	Per cent.	*Per cent.*		*Per cent.*	*Per cent.*
13	2.85	1.02	24	2.67	1.10
14	1.30	.57	25	3.40	1.00
15	1.82	.97	26	2.00	.57
16	1.75	.47	27	3.32	.75
17	2.20	.65	28	2.10	.65
18	5.00	.90	29	2.20	.77
19	1.60	.75	30	1.82	.75
20	2.90	1.07	31	4.32	1.35
21	4.70	.50			
22	2.62	.72	Average	2.79	.84
23	4.51	1.42			

Results of tests of tailings from tables.

Test No.—	Number of tables in use.	Zinc in material fed to tables.	Zinc in tailings from tables.	Test No.—	Number of tables in use.	Zinc in material fed to tables.	Zinc in tailings from tables.
		Per cent.	*Per cent.*			*Per cent.*	*Per cent.*
32	2	2.95	1.35	42	1	9.67	1.92
33	1	6.35	3.75	43	2	5.65	1.62
34	2	4.85	1.70	44	2	4.00	1.10
35	2	4.70	1.55	45	7	4.65	1.42
36	5	4.25	1.57	46	2	15.80	2.30
37	2	3.12	1.00	47	1	6.55	.52
38	2	4.30	1.35	48	1	4.90	1.50
39	3	6.75	1.55				
40	3	3.75	1.20	Average		5.28	1.57
41	2	7.60	1.35				

Generated on 2024-09-28 13:59 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

Results of other tests of tailings from tables.

Test No.—	Number of tables in use.	Zinc in material fed on tables.	Zinc in tailings from separate tables.		
			Coarse.	Medium.	Fine.
		Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
49	6	6.00	1.00	3.12	4.50
50	3	4.80	.25	1.00	2.05
51	3	10.00	2.90	3.50	6.95
52	3	4.30	.52	.92	2.32
53	6	.....	.50 .25	.20	.50
54	4	5.75	.62	.50 1.41	1.35
55	3	12.45	1.75	2.62	4.62
56	3	4.60	1.75	1.05	1.85
57	2	.....	1.20	.97	.....
Average	.....	6.87	1.07	1.53	3.01

Results of tests of material fed to tables.[a]

Test No.—	Zinc in oversize.	Zinc in undersize.	Zinc in tailings from tables.
	Per cent.	*Per cent.*	*Per cent.*
58	3.30	6.35	3.75
59	5.00	4.90	1.50

In test 59 a shaking screen was being used, and in both tests the oversize from 1½-mm. screens was being sent direct to the tailings elevator.

Results of tests of the speed and the stroke of tables.[a]

Test No.—	Table No.—	Speed.	Length of stroke.	Zinc in material fed to tables.	Zinc in tailings from tables.
		Revolutions per minute.	*Inches.*	*Per cent.*	*Per cent.*
62–a	1	2.20	¾	9.25	0.90
62–b	2	1.75	1¼	9.25	3.05
63–a	1	2.44	⅞	7.75	1.25
63–b	2	2.24	½	7.75	3.20

[a] The material fed to both tables was in each case the same; the speeds and strokes of the tables differed.

Losses in different sizes of tailings.

Size of tailings.	Dry weight of sample.	Per cent of total weight.	Quantity of zinc (assay).	Quantity of blende (ZnS) (assay).	Weight of blende content.	Loss.
	Grams.	*Per cent.*	*Per cent.*	*Per cent.*	*Grams.*	*Per cent.*
Oversize to ¾ inch	2.426	15.225	0.77	1.15	27.90	12.256
Size ¾ inch to 3 millimeters	7.579	47.565	.96	1.43	108.38	47.608
Size 3 to 1.5 millimeters	3.817	23.955	.76	1.13	43.13	18.946
Size 1.5 to 0.46 millimeters	1.659	10.412	.71	1.06	17.48	7.678
Undersize through 0.46 millimeter [a]	453	2.843	4.55	6.79	30.76	13.512
Total	15,934	100.000	.96	1.43	227.65	100.000

[a] Ore in overflow water from the large settling tanks (*14*, fig. 4) not included.

Generated on 2024-09-28 13:53 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

Efficiency of concentration in a mill.

Quantity of ore ("dirt") hoisted, tons	326.00
Quantity of ore through mill (5 per cent "culled" and sent to rock pile), tons	309.70
Quantity of zinc concentrates produced, tons	9.235
Assay of zinc concentrates, percentage of zinc	59
Quantity of blende (ZnS, 67 per cent Zn) in concentrates, tons	8.132
Quantity of tailings (309.70—9.235), tons	300.465
Assay of tailings [a] from mill (0.96 of 1 per cent Zn), percentage of ZnS	1.43
Loss of blende (ZnS) in tailings, tons	4.296
Total quantity of blende (ZnS) in ore through mill, tons	12.428
Total recovery of blende (ZnS) from ore through mill, per cent	65.43

LABOR.

Labor conditions in the Joplin district are exceptional. There is perhaps no other district in the United States where more work is accomplished per man. Nearly all the miners and surface men are Americans, and although the wages are high, the labor is unusually efficient.

The underground work at the mines is 8 hours per shift, and with the exception of a few of the larger companies is carried on only in the daytime. The mill shifts, however, are 10 hours each, and when enough ore or "dirt" has been hoisted into the hoppers during the day to warrant a double shift a second mill crew comes on and the mill is kept running until all the ore in the hoppers has been treated. The remainder of the time, if any, is spent in making any necessary repairs or in replacing rolls, so that as little time as possible may be lost by shutdowns during the day shift.

The number of men and the average wage at 57 mines visited, inclusive of all double shifts, are given below:

Number of men and average wages in 57 mines visited.

Designation.	Number of men.	Wages.	
		Total.	Average per day.
Surface laborers.			
Hoist men	124	$333.35	$2.69
Screen men	160	349.35	2.18
Millmen	308	892.34	2.90
Engineers (in power house)	96	301.53	3.14
Shopmen and helpers	135	366.70	2.71
Total surface laborers	823	2,242.92	2.725
Underground laborers.			
Drill men and helpers	796	1,997.35	2.51
Powder men and helpers	49	151.25	3.08
Shovelers	844	[b] 2,447.44	2.90
Tub hookers	100	287.85	2.88
Trackmen	68	178.75	2.63
Extra men (rooftrimmers, pump men, etc.)	197	480.46	2.44
Total underground laborers	2,054	5,543.10	2.70
Total at mines, exclusive of superintendents and managers	2,877	7,786.02	2.706

[a] Ore in overflow water from the large settling tanks (*14*, fig. 4) not included
[b] 4 to 8 cents per tub.

Generated on 2024-09-28 13:56 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

Summary of production and cost of labor.

Total ore hoisted, tons	16,624.5
Average cost of surface labor per ton	$0.135
Average cost of underground labor per ton	.333
Average total labor cost per ton	.468

The above figures show that the average daily wage per man is about $2.70 per day and that there is little difference between the wages for underground and for surface labor.

It has often been stated that this district provides a poor man's camp—that is, that miners with small capital are able to prospect and with a few thousand dollars to start operations. This statement is borne out by many instances in which two or three miners have saved enough money from wages to enable them to enter into partnership and start their own "diggings."

ACCIDENTS AND SAFETY PRECAUTIONS.

The causes of accidents in the Joplin district are numerous, but the most prominent is falls of slabs and bowlders from the roof of the underground workings, other important causes being the misuse of explosives and negligence in the shafts. Although many accidents are unforeseen and unavoidable, there are many due to carelessness on the part of the miners or to lack of proper oversight of the underground workings or lack of discipline over the miners on the part of the ground bosses. According to the twenty-fourth annual report of the bureau of mines, mining, and mine inspection of the State of Missouri for the year 1910, the number of fatal accidents in the mines of the southwestern Missouri district was 32 and of nonfatal 32, whereas for 1909 the number of fatal accidents was 53 and of nonfatal 13. From these figures it can be seen that the number of fatalities decreased considerably during the two years, whereas the number of nonfatal accidents for 1910 was much higher than for 1909. For the year 1910, according to the State report mentioned, 240,418 tons of blende concentrates, 20,959 tons of "silicate," 36,750 tons of lead mineral, 112 tons of "drybone," and 6,200 tons of tripoli were mined, making the total production of mineral during the year 304,439 tons. During this same period 7,887 men were employed in the mines. According to the above figures, there was in 1910 one death per 9,514 tons of mineral produced, or the death rate per 1,000 men employed was 4.05. Much credit, however, is due to the State mine inspectors for increasing diligence and to the mine operators and their employees for a growing spirit of cooperation.

During the past year accidents resulting from falls of roof, the use of explosives, and falls down shafts have occurred. Four shaft accidents resulted fatally. One man was killed by a fall due to his slipping on ice at the mouth of the shaft. Another fell from the tub just before

Generated on 2024-09-28 14:02 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

being lowered. Such accidents could probably have been avoided if substantial railings had been placed around the entrance of the shaft. At most of the mines railings or boards are placed at the mouth of the shaft, but seldom do these extend all the way around, one or two sides being left entirely open. As a man steps out from the tub he takes hold of a near-by post or board nailed to the derrick and pulls himself and the tub to the side of the shaft in order to step out, a procedure that would seemingly result in many accidents. A railing of small iron pipe, placed on each side of the shaft about 2½ to 3 feet above the collar, with an opening of about 2 feet on one side, would enable the men to get a firm grip as they stepped out of the tub.

One rather serious though not fatal accident occurred during the winter of 1911–12, when a man was hit by a piece of ice in the shaft. The shaft had two compartments, so that there was an upward and a downward draft. The water in the down-draft compartment froze as it worked itself down the sides of the shaft, forming large icicles, one of which broke loose and hit one of the men below on the head. To prevent such accidents the first men lowered down the shaft in the morning should break off with their shovels or picks the pieces of ice that have formed on the inner sides of the shaft during the night.

Another cause of accidents in shafts is connected with the hoisting of ore. After the ore has been hoisted up into the derrick, the trap door is lowered to prevent dirt from falling back into the shaft and the tub is dumped on to the grizzly. These doors are made of wood and become worn, so that small pieces of dirt often work through the cracks and fall back down the shaft. The pieces of dirt are usually not large but attain considerable momentum in falling 200 feet and may cause injuries to men below. Also in hoisting ore in tubs the tubs are apt to bump against the sides of the shaft, causing some of the dirt or fairly large pieces of rock to be rolled or jerked out by the impact, with serious results to men below. At a few of the mines injuries of this nature are largely prevented by the installation of an electric bell in the bottom of the shaft, connection being made with the hoisting room. When the hoistman notices that the tub is bumping on the side of the shaft he immediately pushes the bell button at his side, warning the men below to move away from the foot of the shaft. One of the prominent mine managers in the district stated that several serious accidents were prevented each year by this device, which was well worth the cost of installation.

The practice of straddling the side of the tub in riding may result in the miner's hitting his kneecap or foot against the sides of the shaft; consequently all hoist men should be careful not to raise and lower the men at too great a speed.

The number of fatalities underground caused by the use of explosives stands next to that caused by roof falls. It is inexcusable to

Generated on 2024-09-28 13:54 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

allow men to use a steel nail in the end of a loading stick. A spark from the contact of steel and flint while loading holes may cause a very serious accident. Accidents while "squibbing" holes and those due to premature explosion in firing loaded charges occur, and if the machine men do their loading and firing they should be brought to realize the dangers involved.

The fire protection at the mines and mills in general is not of the best. The mills are, as a rule, built of wood, and when a fire once gets started can hardly be extinguished with the fire equipment generally available. Most of the mills are equipped with 50 to 150 feet of fire hose and a few small fire extinguishers, although some have no fire-fighting equipment. In the underground workings there is practically no fire equipment, but this is usually unnecessary, on account of the small amount of timbering done and because of the quantity of water and dampness in most of the mines.

Considering the risks involved, the number of fatalities in the mines of this district is not large, although the death rate should be reduced. This can be accomplished only by more rigid rules in the use of powder and better safeguarding by the management of the men both underground and on the surface.

HEALTH CONDITIONS AT THE MINES.

The health conditions existing at mineral mines in general depend largely upon the material mined, the depth of the deposit, the gases present, and the excellence of the ventilation. The mines in the Joplin district are favorable to sanitary conditions because of the shallow depth of the ore deposits and the lack of injurious gases, such as are found in coal mines and metal mines of great depth. Health considerations, however, have been neglected to a certain extent by many of the operators.

On the whole the circulation of air in the mines is good, a condition largely due to the inspection of the State mine inspectors, who zealously enforce the law compelling operators to sink shafts, drill air holes, or cut drifts to openings, if they are needed, to insure the health of the miners. In a few mines air blowers were used for ventilation, whereas in places in some mines the air was so foul that the light from a carbide lamp was extinguished, but most of these places were abandoned drifts.

Most of the mines in the district, with the exception of the typical "sheet-ground" mines, are rather wet, so that the miners, after the day's work, come from the underground workings with wet shoes and clothing. It is here that conditions could be considerably bettered, for there are few mines that have special houses equipped with lockers where the men can change their clothing before leaving the mine. It is true that many of the miners do not live far from the

Generated on 2024-09-28 14:00 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

mines and would prefer to hurry home to change. However, a large number of miners have rather a long distance to go and are obliged to drive or to use the electric railway lines. Where there is no place at the mines to change their clothing, they are subjected to a sudden change of air and temperature at the surface, especially if they drive home, and in consequence are constantly running the chance of taking severe colds with serious results. Although the men become hardened to these conditions and can usually resist any temporary cold, the constant exposure day after day gradually reduces their vitality, and any precautions taken to prevent such conditions are highly desirable. At one of the mines a newly built changing house was equipped, not only with lockers but also with a large wash sink and a place for hanging and drying wet clothes. The manager of this mine stated that he never had any trouble in obtaining men to work for him. Equipment such as that mentioned would not only benefit the health of the miners but should also result in increased efficiency on their part. Greater interest in their health and welfare on the part of mine management would also help to insure the mines against labor trouble.

In the mines that are practically free from water and dampness there is usually more or less dust from the drilling and blasting. This dust may be carried in the air and inhaled by the miners, causing lung trouble if breathed for a considerable time, just as lead poisoning is brought about by breathing the fumes at lead smelters. The extent of harm done by inhaling the dust in metal mines is not definitely known, but it is believed that much of the lung trouble often known as "miners' consumption" has been caused by it. Some of the operators in the district lessen the injurious effect of rock dust by dampening the faces, but this is done at few mines. This wetting of the face, however, should be done at all mines where rock dust is prevalent, for it would without doubt prevent much sickness and spreading of disease. In this connection it may be stated that the Bureau of Mines, in cooperation with the Bureau of the Public Health, is to investigate a number of problems relating to the hygiene of mines and to the mine industry, the investigations being conducted with regard to scientific as well as administrative problems and including a study of the prevalence and prevention of lung diseases among miners.

CONCLUSION.

Taking the Joplin district as a whole, the methods of mining and milling are by no means perfect, although many of the mines and concentrating plants are obtaining excellent results under the existing conditions. There are, however, many mines where improvements can be made, or where greater efficiency or a higher recovery

Generated on 2024-09-28 13:59 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

can be realized. Many of the operators should make more definite endeavor to determine where the losses are greatest by conducting careful tests more frequently, as the results for one week are not necessarily representative for the weeks or months following. A relation frequently existing is that as the ore in the mines becomes richer, the efforts to save all the mineral possible become more lax.

The only sorting worthy of mention is the culling of large bowlders of waste rock. Because of the large quantity of waste rock present, especially in the "hard-ground" mines where there is often a comparatively thick layer of barren rock between two or more "runs" of ore, it is reasonable to assume that some system of hand sorting, underground or in the mill, could be used successfully. The discarding of a larger percentage of waste rock would result in a higher metal recovery in the mills and would reduce the amount of wear and tear on the machinery per ton of ore mined.

There are other features besides efficiency in the mining and concentration of the ores that can be improved. Safety precautions for the prevention of accidents in the mines, and improvements in the conditions affecting the health of the miners, are no less important, and in some of the mines should have more attention from the operators than at present. Such subjects as the use of explosives, mine timbering, fire protection, and safety appliances require careful study and should be just as vital to the mine operator as the number of tons of ore mined and treated during the day's run.

The tendency is to sacrifice efficiency for capacity, but many of the operators are almost obliged to make this sacrifice because of the high royalties that they have to pay. The landowners seldom mine their own properties, but lease them for royalties of 10 to 15 per cent of the gross output of mineral. The first lessees sublease the properties, increasing the royalties to 20 or even 30 per cent. The operators, by reason of these heavy royalties, are compelled to use methods by which the largest capacity possible can be obtained. Consequently, in order to make any profit at all, they often greatly overcrowd the mills. Again, some of the operators and miners try to mine only the higher grade ore, leaving much of the lower grade in the mine, or in other words, it is often the practice to gouge the ore where the royalties are excessive. On the other hand, the system of leasing has its advantages. The properties of the landowners are constantly being prospected for rich ore bodies, and the district is not only alive, but to a certain extent is free from labor difficulties. A system of royalties based on the net profits, or reduced royalties under the present system, would in many of the mines result in more efficient mining. A closer mill saving would also be possible, the life of the mines would be lengthened and the general conditions in the district would be improved in many ways.

Generated on 2024-09-28 13:55 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF MINING.

The following Bureau of Mines publications may be obtained free by applying to the Director, Bureau of Mines, Washington, D. C.

BULLETIN 10. The use of permissible explosives, by J. J. Rutledge and Clarence Hall. 1912. 34 pp., 5 pls., 4 figs.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larson, Frank Haas, and Carl Scholz. 204 pp., 14 pls., 28 figs. Reprint of United States Geological Survey Bulletin 425.

BULLETIN 44. First national mine-safety demonstration, Pittsburgh, Pa., October 30 and 31, 1911, by H. M. Wilson and A. H. Fay, with a chapter on the explosion at the experimental mine, by G. S. Rice. 1912. 75 pp., 7 pls., 4 figs.

BULLETIN 45. Sand available for filling mine workings in the northern anthracite coal basin of Pennsylvania, by N. H. Darton. 1912. 33 pp., 8 pls., 5 figs.

BULLETIN 46. An investigation of explosion-proof mine motors, by H. H. Clark. 1912. 44 pp., 6 pls., 14 figs.

BULLETIN 48. The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell. 1913. 50 pp., 3 pls., 7 figs.

BULLETIN 52. Ignition of mine gases by the filaments of incandescent electric lamps, by H. H. Clark and L. C. Ilsley. 1913. 31 pp., 6 pls., 2 figs.

BULLETIN 56. First series of coal-dust tests in the experimental mine, by G. S. Rice, L. M. Jones, J. K. Clement, and W. L. Egy. 1913. 115 pp., 12 pls., 28 figs.

BULLETIN 60. Hydraulic mine filling; its use in the Pennsylvania anthracite fields; a preliminary report, by Charles Enzian. 1913. 77 pp., 3 pls., 13 figs.

TECHNICAL PAPER 11. The use of mice and birds for detecting carbon monoxide after mine fires and explosions, by G. A. Burrell. 1912. 15 pp.

TECHNICAL PAPER 13. Gas analysis as an aid in fighting mine fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp., 1 fig.

TECHNICAL PAPER 19. The factor of safety in mine electrical installations, by H. H. Clark. 1912. 14 pp.

TECHNICAL PAPER 21. The prevention of mine explosions; report and recommendations, by Victor Watteyne, Carl Meissner, and Arthur Desborough. 12 pp. Reprint of United States Geological Survey Bulletin 369.

TECHNICAL PAPER 22. Electrical symbols for mine maps, by H. H. Clark. 1912. 11 pp., 8 figs.

TECHNICAL PAPER 23. Ignition of mine gas by miniature electric lamps, by H. H. Clark. 1912. 5 pp.

TECHNICAL PAPER 24. Mine fires, a preliminary study, by G. S. Rice. 1912. 51 pp., 1 fig.

TECHNICAL PAPER 28. Ignition of mine gas by standard incandescent lamps, by H. H. Clark. 1912. 6 pp.

TECHNICAL PAPER 29. Training with mine-rescue breathing apparatus, by J. W. Paul. 1912. 16 pp.

TECHNICAL PAPER 32. The cementing process of excluding water from oil wells, as practiced in California, by Ralph Arnold and V. R. Garfias. 1913. 12 pp., 1 fig.

TECHNICAL PAPER 40. Metal-mine accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 54 pp.

TECHNICAL PAPER 44. Safety electric switches for mines, by H. H. Clark. 1913. 8 pp.

Generated on 2024-09-28 14:02 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google
Digitized by Google
Original from PRINCETON UNIVERSITY

TECHNICAL PAPER 46. Quarry accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 32 pp.

TECHNICAL PAPER 47. Portable electric mine lamps, by H. H. Clark. 1913. 13 pp.

TECHNICAL PAPER 48. Coal-mine accidents in the United States, 1896–1912, with monthly statistics for 1912, compiled by F. W. Horton. 1913. 74 pp., 10 figs.

TECHNICAL PAPER 53. Proposed regulations for the drilling of oil and gas wells, with comments thereon, by O. P. Hood and A. G. Heggem. 1913. 28 pp., 2 figs.

MINERS' CIRCULAR 3. Coal-dust explosions, by G. S. Rice. 1911. 22 pp.

MINERS' CIRCULAR 4. The use and care of mine-rescue breathing apparatus, by J. W. Paul. 1911. 24 pp., 5 figs.

MINERS' CIRCULAR 5. Electrical accidents in mines; their causes and prevention, by H. H. Clark, W. D. Roberts, L. C. Ilsley, and H. F. Randolph. 1911. 10 pp., 3 pls.

MINERS' CIRCULAR 6. Permissible explosives tested prior to January 1, 1912, and precautions to be taken in their use, by Clarence Hall. 1912. 20 pp.

MINERS' CIRCULAR 9. Accidents from falls of roof and coal, by G. S. Rice. 1912. 16 pp.

MINERS' CIRCULAR 10. Mine fires and how to fight them, by J. W. Paul. 1912. 14 pp.

MINERS' CIRCULAR 11. Accidents from mine cars and locomotives, by L. M. Jones. 1912. 16 pp.

MINERS' CIRCULAR 12. The use and care of miners' safety lamps, by J. W. Paul. 1913. 16 pp., 4 figs.

Generated on 2024-09-28 14:00 GMT / https://hdl.handle.net/2027/njp.32101054990351
Public Domain, Google-digitized / http://www.hathitrust.org/access_use#pd-google

Digitized by Google

Original from
PRINCETON UNIVERSITY